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MINING METHODS AND COSTS AT THE NEW CORNELIA BRANCH, PHELPS DODGE CORPORATION, AJO, ARIZ.

BY

GEORGE R. INGHAM AND ALFRED T. BARR

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MINING METHODS AND COSTS AT THE NEW CORNELIA BRANCH, PHELPS DODGE CORPORATION, AJO, ARIZ.[1]

By George R. Ingham[2] and Alfred T. Barr[3]

INTRODUCTION

This paper is one of a series being prepared by the Bureau of Mines on mining methods and costs at various mines in the United States, and is a description of the mining practice at Ajo, Ariz., where the New Cornelia Branch of the Phelps Dodge Corporation mines copper ore from an open pit, and operates a concentrator for the treatment of the ore thus mined.

ACKNOWLEDGMENTS

The authors wish to express their appreciation to M. Curley, manager of the New Cornelia Branch of the Phelps Dodge Corporation, for his review of this paper and permission to publish it. They also wish to acknowledge the assistance given by W. C. Lawson, assistant engineer, in the preparation of the paper.

HISTORY

The copper deposit at Ajo was one of the first to be discovered in the Southwest, but its development into a successful mining enterprise occurred at a comparatively recent date. For a period of nearly a half-century after the discovery of copper in the district, mining was confined to small bodies of high-grade ore which were found on the edge of what was later proved to be a large deposit of low-grade copper ore. In the sixties, native copper and cuprite ore were mined from shallow surface workings and hauled by teams to San Diego and later to Yuma, and from these ports transported in sailing ships to Swansea, Wales. After the Southern Pacific Railroad was built, ore was hauled 43 miles to Gila Bend, which was the nearest shipping point on the railroad. The mining of small high-grade orebodies continued in an intermittent way until the panic of 1907, when all properties were closed down, and the camp was practically deserted. It is probable that at no time were these operations very profitable, the chief factors against their success being the lack of water in the district, the long haul to a seaport or railroad, and the high cost of mining the small bodies of ore. During this period practically no work had been done on the great carbonate outcrop of the main orebody; several years elapsed after the suspension of operations in 1907 before the potential worth of the deposit was realized.

The first real attempt to explore the orebody was made in the winter of 1909-1910. At that time the major portion of the mineralized area was held by two companies, the New Cornelia Copper Co. and the Rendall Ore Reduction Co. The Lewisohn interests secured an option

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2 - One of the consulting engineers, U. S. Bureau of Mines, and mine superintendent, New Cornelia Branch, Phelps Dodge Corporation.

3 - One of the consulting engineers, U. S. Bureau of Mines, and chief engineer, New Cornelia Branch, Phelps Dodge Corp.

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on the New Cornelia property and put down five diamond-drill holes around the base of Copper Mountain, the most prominent outcrop in the basin. Although three of these holes cut material which is now considered ore of commercial grade, it was not considered so at that time, and the option on the property was given up. Likewise, Seeley W. Mudd and associates secured an option on the Rendall property, along the southern and eastern edges of the main orebody, but after drilling six holes, they gave up the option. Four of the holes were blanks and two cut ore averaging about 1-1/4 per cent copper.

Despite the disappointing results of these first attempts at drilling the deposit, the Calumet & Arizona Mining Co. took an option on all available stock of the New Cornelia Copper Co., and in December, 1911, started a drilling campaign, which eventually developed a considerable tonnage of low-grade ore. The first hole was drilled on the west side of Copper Mountain, 200 feet north of the best hole drilled by the Lewisohn company, and over 200 feet of ore was cut which averaged about 1-1/2 per cent copper. As the results of this hole were encouraging, drilling was continued until an area of nearly 80 acres had been explored, and 40,000,000 tons of ore averaging 1-1/2 per cent copper had been developed.

In the meantime, the Rendall property had been acquired and developed by James Phillips, jr., and James P. Gaskill, under the name of the Ajo Consolidated Copper Co. The ground lying between the Rendall and the New Cornelia properties, known as the Childs group, was taken under option by representatives of the U. S. Smelting, Refining & Mining Co., but was given up after a few churn-drill holes had been put down. The New Cornelia Copper Co. afterwards acquired the holdings of the Ajo Consolidated Copper Co., the Childs group of claims, and a large block of ground north of the original holdings, on which the townsite, tailings dumps, and plants are now located.

The orebody which was being developed at Ajo was different in many respects from other so-called "porphyry coppers" in the Southwest, and it was evident that some departures from the usual methods of mining and treatment would be necessary to insure the success of the enterprise. The upper part of the orebody consisted of oxidized or carbonate ore, which extended from the surface down to an average depth of more than 50 feet. This ore had about the same copper content as the sulphides, which lay immediately underneath, but at that time no process for the treatment of such low-grade oxidized ore was in use. As it seemed that economical mining of the orebody meant the use of open-pit rather than underground methods, a successful method of treating the carbonate ore, which had to be mined to uncover the sulphide ore, was necessary in order to bring the property to an early stage of production and to lessen the expense of stripping the sulphide ore.

Soon after development of the orebody was under way, experiments were being made to determine a method of treating the carbonate ore. It was found that the ore could be successfully leached with sulphuric acid, and the copper which went into solution could be deposited on sheets by electrolysis. After three and a half years of experimental work, during which about 15,000 tons of carbonate ore was treated in pilot plants, a commercial plant with a capacity of 5,000 tons daily was built. This plant was put into operation in May, 1917, and continued in operation until July, 1930, when practically all of the carbonate ore had been mined.

By 1923 sufficient sulphide ore had been uncovered to warrant building a concentrating plant for its treatment. The original plant, which had a rated capacity of 5,000 tons daily, was put in operation in January, 1924. A year or so before the carbonate ore was exhausted, the concentrator was enlarged and improved to give a capacity of approximately 15,000 tons daily.

Before construction work on the leaching plant was started, the Tucson, Cornelia & Gila Bend Railroad had been completed from Gila Bend to Ajo, and an adequate water supply for the subsequent operations had been developed in the valley 7 miles north of Ajo.

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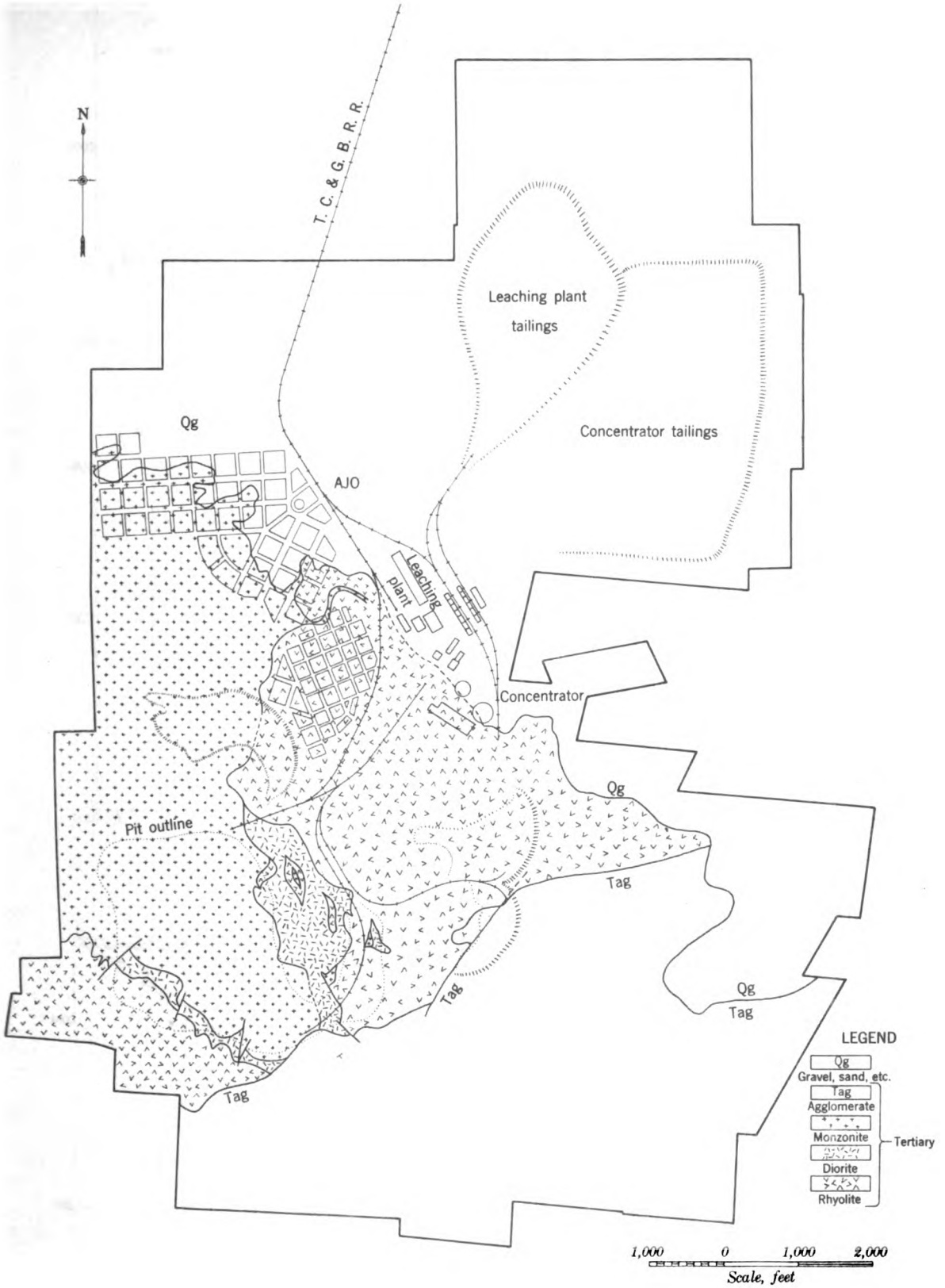

Figure 1.—Geologic map of the New Cornelia property

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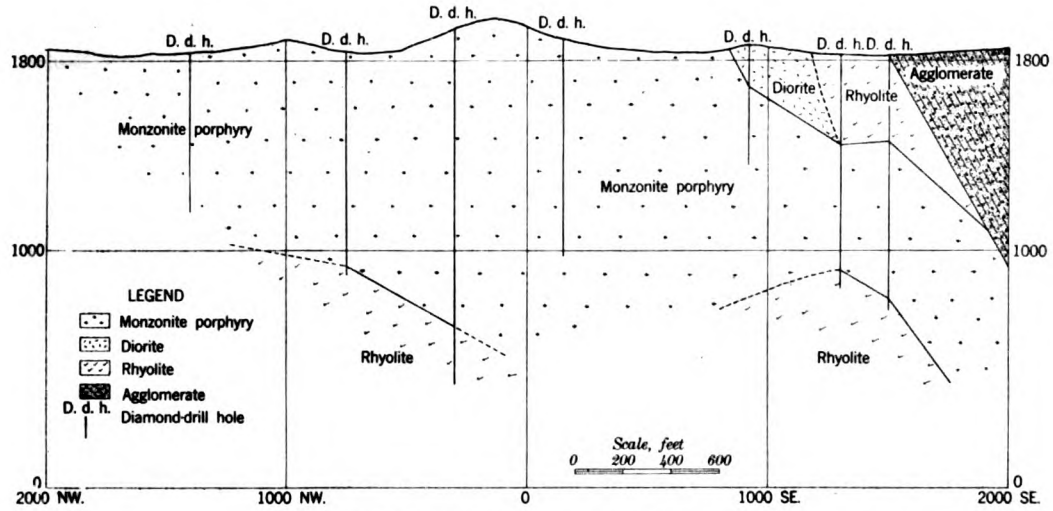

Figure 2.—Geologic section through orebody, looking northeast

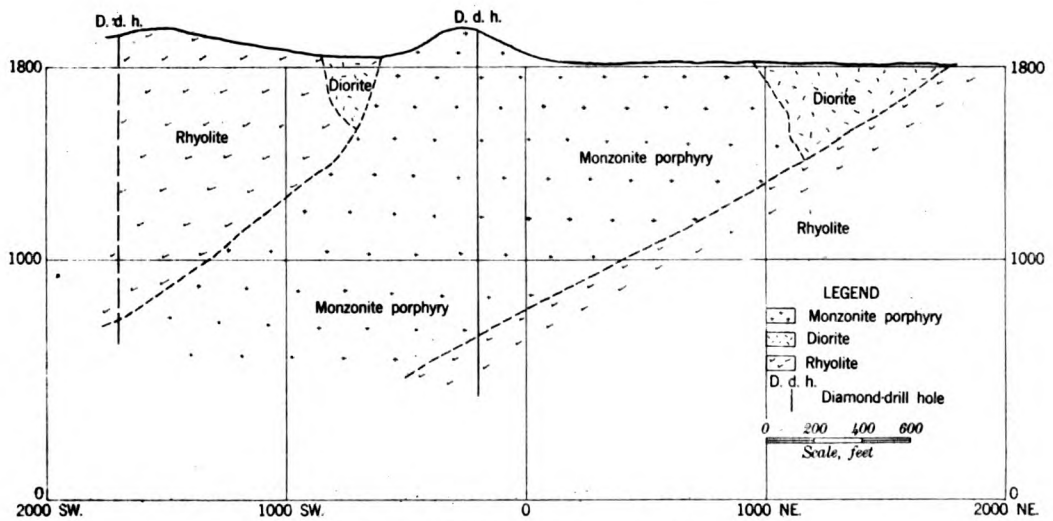

Figure 3.—Geologic section through orebody, looking northwest

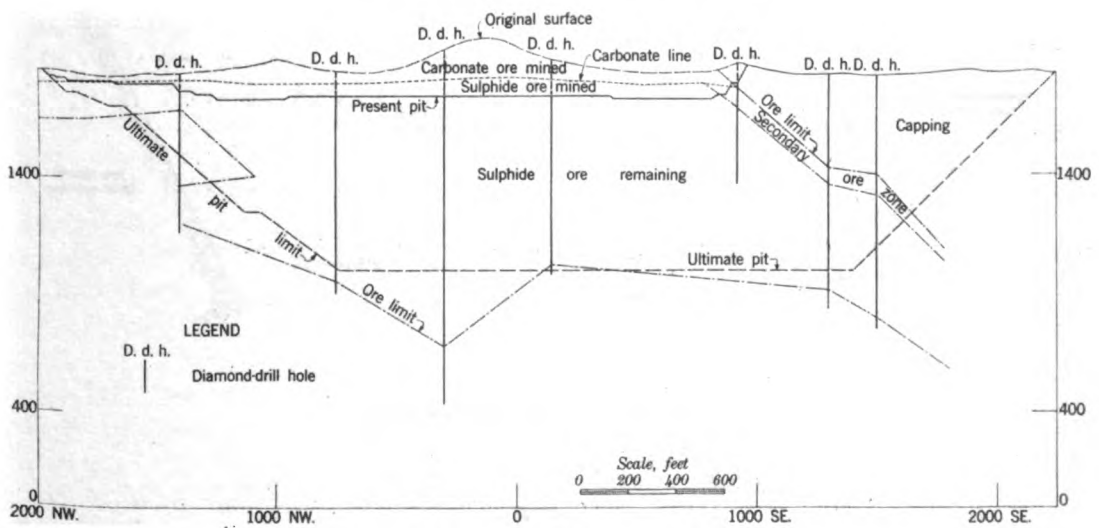

Figure 4.—Typical section of orebody, looking northeast

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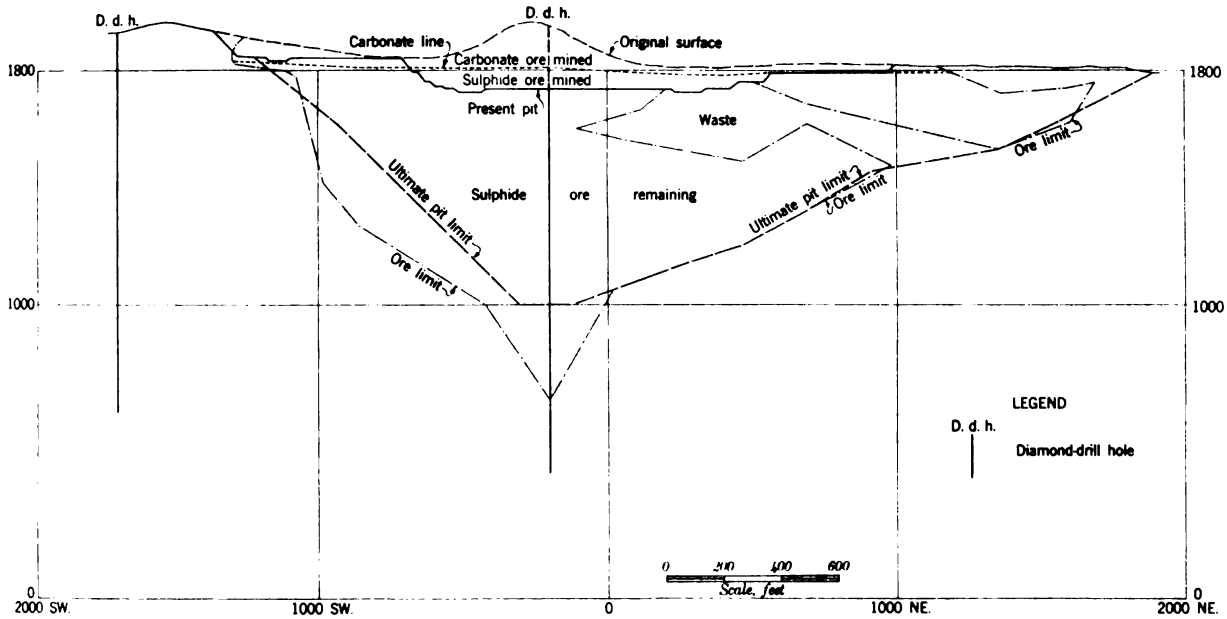

Figure 5.—Typical section looking northwest

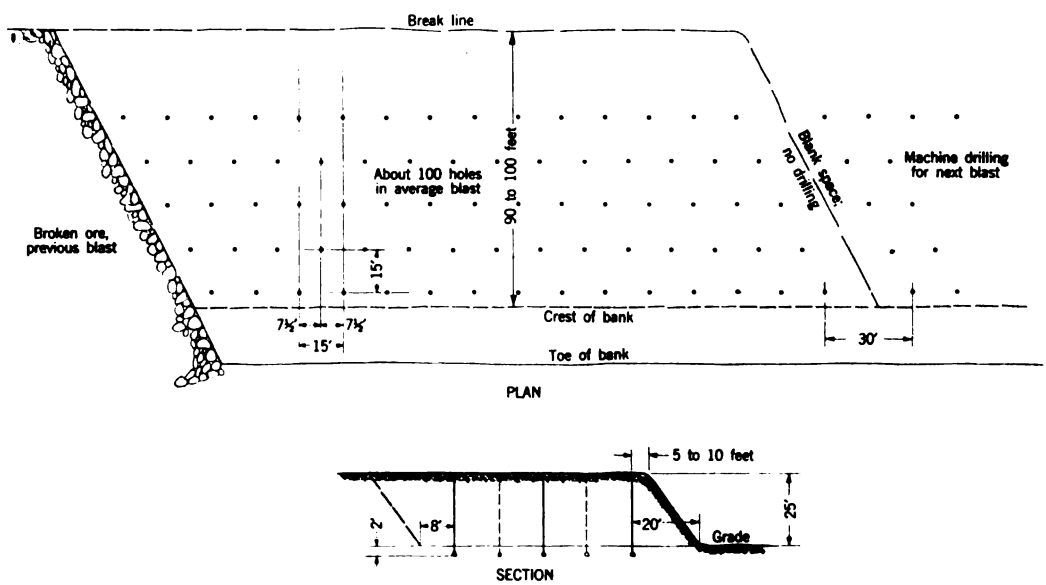

Figure 6.—Sketch of standard blast

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In 1929 the New Cornelia Copper Co. was consolidated with the Calumet & Arizona Mining Co., and in 1931 this company consolidated with the Phelps Dodge Corporation. Since October 1, 1931, the New Cornelia property has been operated under the name of the New Cornelia Branch of the Phelps Dodge Corporation.

GEOLOGY

The orebody is located in the southern end of a large laccolith of monzonite and monzonite porphyry (fig. 1) which has intruded andesite and rhyolite flows, and which has, according to Joralemon,[4] uplifted the rhyolite beds to form a dome; the crest of the dome has been eroded away in fairly recent times. The exposed portion of the monzonite has a length of about 2 miles, and a width of over a mile at its northern end and about 1,000 feet at its southern end. The southern tip of the monzonite dips steeply toward the south and southeast, and is covered with rhyolite and beds of recent agglomerate (figs. 2 and 3). The northern end of the intrusive is covered with alluvium and recent lava flows. Bordering the southern point of the monzonite intrusion, there is a strip of diorite from 200 to 1,000 feet wide, which is presumably an earlier intrusion than the monzonite.

The orebody is composed of monzonite and diorite, and to a lesser extent rhyolite, which have been mineralized by copper-bearing solutions. Copper minerals have been deposited along fracture planes, and disseminated throughout the rock mass to form a large body of low-grade copper ore. During the primary mineralization, chalcopyrite, bornite, and some pyrite were deposited; then, after the capping of the orebody was eroded away and the top of the ore was exposed to oxidizing agencies, a blanket of oxidized ore was formed covering the primary ore. Secondary enrichment played only a minor part in the formation of the greater portion of the orebody, with the result that oxidized ore of nearly the same grade as the underlying primary sulphide ore was formed. Some secondary sulphide enrichment took place along the southeastern edge of the orebody, and here the primary ore is capped with chalcocite ore which is somewhat higher in copper than the main body of the oxide and sulphide ore.

The outcropping of the orebody is roughly circular in shape with a diameter of about 2/3 mile. The average thickness of the orebody, including ore which has been mined, is 425 feet, and the maximum thickness nearly 1,000 feet (fig. 4). Oxidized ore extended from the surface down to an average depth of 55 feet, and the depth varied from 20 to 190 feet; the contact between the oxidized ore and the underlying sulphide was, broadly speaking, nearly a level plane, but local variations of as much as 50 feet were encountered. The principal copper minerals in the oxidized ore are malachite and azurite, the former being the more prevalent. Chalcopyrite and bornite are the principal ore minerals in the primary sulphide ore, a greater proportion of the copper being in the form of chalcopyrite. The secondary ore zone forms a crescent about the southern edge of the orebody, and continues down on the flank of the monzonite under barren rhyolite and agglomerate. This secondary ore averages about 50 feet in thickness; chalcocite, in veinlets and finely disseminated throughout the rock, is the principal copper mineral. Other ore minerals which are found in the orebody, but not abundantly, are tetrahedrite, native copper, cuprite, tenorite, and chrysocolla.

PHYSICAL CHARACTERISTICS OF ORE AND ENCLOSING ROCKS

A typical cross section of the orebody (fig. 5) shows the sides sloping downward toward a definite keel of deep ore along the bottom. This keel, which is along the longitudinal

4 – Joralemon, Ira B., The Ajo Copper Mining District: Trans. Am. Inst. Min. Eng., vol. 49, 1914, p. 593.

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axis of the orebody, strikes about north 30° west and is fairly flat until it plunges downward near its south end (fig. 4). With the exception of the southern portion of the orebody, its shape is such that most of the ore can be mined by open-pit methods with the removal of a relatively small proportion of waste. To strip and mine the ore from the south end of the orebody, an increasing proportion of waste will be handled until the ratio of waste to ore will be such that an underground method of mining will be cheaper.

Strictly speaking, the grade of the ore, as shown by assays of drill hole samples, is irregular; however, the grade of the ore treated is easily kept uniform by mixing ores from various sections of the pit. The average grade of the sulphide ore produced during the year 1930 was 1.24 per cent copper, and the highest and lowest monthly grades during that year were 1.41 and 1.12 per cent copper, respectively.

The ore is on the average very hard, and breaks into large bowlders on account of the presence of widely spaced fracture planes throughout the rock. The central portion and west side of the orebody are highly siliceous, and present a particularly hard drilling and blasting problem. A few minor faults occur within the orebody, but none have been encountered which have seriously affected mining operations. In general, the waste and overburden material is softer than the ore.

METHODS OF PROSPECTING AND EXPLORATION

Drilling

The orebody has been developed almost exclusively by means of diamond drilling. Out of a total of 250 holes drilled on the property, 236 are diamond-drill holes with a total footage of 121,900 feet. The remainder, or 14 holes, were drilled with churn drills, and have a total footage of 4,849 feet. Prior to 1926 most of the holes were drilled to relatively shallow depths; the average depth of 178 diamond-drill holes and the 14 churn-drill holes was only 365 feet. Many of these holes were stopped in ore, or at least in material which was later recognized as ore in view of the improved costs and metallurgical results obtained in the mining and concentration of the sulphide ore; in order to develop the probable extension of the orebody both in depth and laterally, a drilling campaign was started in April, 1926. During this campaign, 58 holes, with an average depth of 976 feet, were drilled.

The majority of the holes in the orebody were drilled vertically downward at the intersections of north-south and east-west coordinates at 200-foot intervals, but in the recent drilling of the deeper ore in the main orebody, 400 feet was adopted as the standard interval between holes. All of the holes were drilled with an A bit, with the exception of a few holes put down with an E bit. The average core recovery for all diamond drilling to date (March, 1931) is 49.8 per cent, with a maximum recovery for any individual hole of 78.4 per cent, and a minimum of 8.3 per cent. Core recoveries above the average were obtained in holes drilled in the western and southern parts of the orebody, while drill holes in the eastern portion of the main orebody gave core recoveries below the average. A greater percentage of core was recovered from deep holes than from shallow holes. Of the total footage of diamond drilling, 62.4 per cent was in monzonite, 21.9 per cent in rhyolite, 7.5 per cent in diorite, 7.2 per cent in agglomerate, and 1 per cent in other formations, mostly small dikes cutting the orebody.

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Test Pitting

During the period of the early development of the orebody by diamond drilling, test pits were sunk on drill holes in order to check the results of the drill hole samples; later pits were sunk on drill hole locations in advance of drilling, in order to expedite the development program. Seventy-seven test pits were sunk with a total footage of 3,955 feet, of which 1,234 feet was for checking diamond drilling, and 2,721 feet was in advance of drilling. During the sinking of these test pits, every tenth bucket of broken material windlassed from the hole was taken for a sample. After the test pits were finished, they were sampled on all four sides by means of vertical channels cut 6 inches wide by 3 inches deep. The channel samples averaged 0.15 per cent lower in copper than the bucket samples, and slightly lower than the diamond-drill hole samples.

Drifting and Raising

As most of the test pitting was done in carbonate ore, drifts were run on coordinate lines from the bottom of two of the deepest test pits to check more thoroughly the results of the drilling in sulphide ore. Raises were also put up from the drifts at points where they cut drill holes. The total amount of drifting was 1,513 feet, and of raising, 142 feet.

SAMPLING AND ESTIMATION OF TONNAGES AND VALUES

Drill-Hole Sampling and Calculation of Sample Values

All diamond drilling to date has been done under contract, but the sampling has been done by representatives of the company. The sampling crew consists of a head sampler, and one sampler for each drill shift; the work is under the supervision of the chief engineer. The head sampler directs the work of the samplers at the drills, splits the cores, and makes daily reports to the chief engineer. The samplers are constantly on duty at the drills while drilling is in progress, and attend to the collection of the sludge, drying and sacking of the sludge samples, and measuring and sacking of the cores. With the exception of drilling done in barren capping, samples are taken at intervals of 5 feet. All of the sludge from a run is caught in cylindrical galvanized iron tanks, which are 30 inches in diameter and 26 inches high. Enough tanks are kept at each drill to hold all of the sludge from one run and to give at least one extra tank for starting the next run. Usually five tanks give ample capacity, and when drilling in material which settles quickly four tanks are sufficient. The sludge is allowed to settle as long as practicable, clear water is then siphoned from the tanks, and the thickened sludge is transferred to tubs for drying. Drying is done over a wood fire, but care is taken not to heat the sludge enough to roast the sulphide minerals. After the sludge is dried it is sacked and sent to the assaying laboratory, where it is mixed and samples are cut out for assay. A small sample is returned to the engineering department for filing.

The core which is recovered from a run is measured at the drill and sacked. It is then taken to the sample house, where it is weighed to check the measured lengths, and split lengthwise to give two equal portions of core, one of which is sent to the laboratory for assay and the other retained for geologic study or for a check sample if necessary. A small specimen from the reject half of the core is marked with the hole number and depth and filed for ready geologic reference. These specimens are from 1-1/2 to 2 inches long, and 20 of them, representing 100 feet of drilling, are filed together in one box.

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The head sampler makes out a report for each hole, showing the progress for the day, the length of core, rock formation, ore minerals, and hardness of rock for each 5-foot run. These reports are used in conjunction with the assays of the core and sludge reported by the assaying laboratory, to make out the complete logs of the holes. The core and sludge assays are combined according to the relative weight of the core recovered and the calculated weight of sludge on the basis of total sludge recovery. A table showing factors for core lengths from 1 to 60 inches is used to facilitate calculation of the combined assay. Composite samples from every 50 feet of hole are analyzed for iron, alumina, silica, calcium oxide, and sulphur, and assayed for gold and silver. The analyses of the composites are also entered on the drill-hole log.

Test-Pit Sampling

The method of sampling test pits by channels has been discussed under "Prospecting and Exploration."

"Air-Drill" Sampling

Small samples composed of blast-hole drill cuttings are used in part to control production. The cuttings from a blasting hole which is being sampled are caught on a discarded filter blanket placed around the collar of the hole. A galvanized-iron cone about 2 feet in diameter with a hole in the top for the passage of drill steel stops the cuttings when they are blown from the hole and retains them on the blanket. This method of sampling is inexpensive, and can be used without any hindrance or delay to mining operations. This sampling is further discussed under "Estimation."

Estimation of Tonnages and Grades

Horizontal sections at 25-foot intervals, corresponding to present or future mining levels, are used for estimating tonnages and grades of open pit ore. All holes are plotted on the sections which they pierce, and the assays between levels are averaged for all holes and these average assays marked on the section under the hole number. The figures obtained by combining the core and sludge assays are used without further correction. Ore limits and pit limits are drawn on the section and the areas governed by the grades of the various holes are outlined. Each area is then measured with a planimeter which is set so that tonnages are read direct; the tonnage is multiplied by its grade for grade units; the i dividual tonnages and grade units are summed for the level tonnage and grade; and finally, the level tonnages and grade units are summed for total open-pit tonnage and grade. A factor of 12.5 cubic feet per ton is used for conversion of volumes of rock in place to tonnages.

The foregoing method of estimation is not applicable to the southern and southeastern portion of the orebody, which dips steeply toward the south and southeast. This ore is divided into vertical triangular prisms, the tonnage and grade of each prism is calculated, and the sum of the triangular prisms gives the total tonnage.

Diamond drill hole assays are used in estimates for planning mine operations entailing relatively large tonnages, but are not used extensively in the control of the daily production of ore. Plans for the production of ore for a year, or for a longer period are based entirely on estimates from diamond drill hole assays.

The daily ore production is controlled by the assay of samples, called "air-drill samples," taken from the cuttings of the blasting holes in the benches which are being mined. Monthly plans are made from estimates, using the air-drill assays in conjunction with diamond drill hole assays.

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From one to four blasting holes, the number depending on the width of the blast, are sampled at intervals of 20 feet along each bank being drilled. The holes that are sampled are located with a transit and the corresponding assays plotted on the monthly pit map from which daily and monthly ore estimates can be made. The locations of the air-drill holes sampled are marked in the pit with stakes set on the loading tracks opposite the proper lines of holes. Corresponding assays of these samples are on file at the mine office; the loading of ore and waste is controlled by these assays. In addition a daily estimate is made by the engineering department of the ore production from each shovel by using the air-drill samples and the number of cars loaded by the shovel. The mine department each day furnishes the engineering department with the locations of shovels with reference to the sample stakes and the number of cars of ore and waste which were loaded during the previous day's work. The grade of ore obtained by this method of sampling and estimation checks very closely with the grade obtained from the automatic sampling at the concentrator. In the last six months the actual grade of ore, according to automatic samples taken at the concentrator, was slightly less than 1/10 of 1 per cent of copper higher than the grade obtained by air drill hole samples.

CHOICE OF MINING METHOD

Open-pit mining with power shovels is particularly well adapted to the major portion of the New Cornelia orebody. On account of the low grade of the ore, it was necessary to use a method of mining by which the ore could be mined very cheaply. The hardness of the ore alone was enough to decide against caving methods, and the lack of an appreciable amount of overburden made open-pit mining especially attractive. The orebody, being mostly of primary origin, is not as uniform in grade as some of the low-grade porphyry deposits of secondary origin, and consequently a method of mining which permits the separation of ore and waste is particularly desirable. Material below the minimum ore grade can easily be separated in open-pit mining. Plans have been made to mine at least 80 per cent of the developed orebody by open-pit methods, but eventually underground methods will be used to mine a portion of the orebody which has excessive overburden.

In determining the limiting plane between open-pit ore and underground ore, the relative cost of mining by the two methods is the first consideration. For instance, if the cost of mining a ton of ore by underground methods is four times the cost of mining a ton of material (ore or waste) by open-pit methods, a stripping ratio of 3 of waste to 1 of ore could be used in determining the cut-off between underground and open-pit ore. The cost of steam-shovel mining in the New Cornelia pit has been determined by many years of operation, but, as no large-scale underground mining has as yet been attempted, this cost can only be approximated by estimates. A stripping ratio of 3 to 1 has been used in determining the ultimate limits of a pit in places where it was necessary to differentiate between underground and open-pit ore, for the purpose of estimating ore reserves. The pit has at no place approached the stage where in actual mining operations it was necessary to decide between the two methods of mining. Other factors than the relative costs of open-pit and underground mining will also have a bearing on the problem. Lack of dilution of the ore, or separation of waste from ore, flexibility of operation, a greater recovery of ore, and familiarity through years of experience with open-pit methods will be factors in deciding how far power-shovel mining can be carried. A more pertinent problem than that of differentiating between underground ore and open-pit ore is that of determining the limits of the pit in material that can be mined by open-pit methods only, or, in other words, determining the minimum grade of ore that can be mined at a profit. In this case the controlling factors are the total cost of mining and treating a ton of ore of a certain grade, and the value of the copper which can be recovered

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from the ore. The total cost of mining and treating a ton of ore will vary according to the grade of the ore for the reason that smelting, refining, and other expenses are dependent upon the metal content of the ore, whereas only mining, transportation of ore, crushing and conveying, and milling expenses are constant for all grades of ore. With the unit costs for these different items, the total costs for various grades of ore can readily be calculated and compared with the amounts that would be received from the sale of the recovered copper at various prices for that metal. The minimum grade of the ore that is to be included in the pit should show a fair operating profit, but material that is mixed with ore above the minimum grade and must be mined in the process of mining the higher grade ore may be treated even though it shows no real profit. In this case, however, milling capacity is the important consideration, for ore that can be treated at a bare profit should not exclude a like amount of ore that would give a fair profit.

MINING METHOD[5]

The original diamond drilling of the orebody which was started in December, 1911, and finished in May, 1913, developed 12,000,000 tons of carbonate ore averaging 1.54 per cent of copper, and 28,000,000 tons of sulphide ore averaging 1.50 per cent of copper. The removal of the carbonate capping from the sulphide ore was the first consideration in the mining of the orebody. Fortunately, it was found that the carbonate ore could be treated profitably by leaching with sulphuric acid solution, followed by precipitation of the copper by electrolysis, so that what might have been purely a stripping problem was turned into a profitable operation. Little or no waste overburden occurred over the carbonate ore. The mining of three carbonate hills, rising with steep slopes to elevations from 115 to 165 feet above the general surface, was the most difficult part of the operation. It was finally decided to attack the hills from their bases and mine them in one lift rather than to try to establish benches at intermediate levels between their bases and crests. The maximum height of bank thus encountered was 160 feet.

Both coyote and churn drill hole blasting were used in breaking the ore in the hills. It was found that coyote blasting was more economical for high banks, over 45 feet, whereas churn drilling was better adapted to low banks. Piston air drills eventually superseded churn drills for drilling banks under 30 feet in height, and these in turn have been replaced by hammer machines. The largest single blast, in which one tunnel and 10 drill holes were shot, broke 285,000 tons of ore.

The broken ore was loaded by means of steam shovels into 20-yard, automatic, air-dump cars and hauled over standard-gage tracks with steam locomotives to the crushing plant. Five or six cars normally constituted a train and the average length of haul was 1 mile.

After the hills had been cut down, mining levels at vertical intervals of approximately 30 feet were established for the purpose of mining the balance of the carbonate ore, but as the contact between the carbonate and sulphide ore was a warped surface, with as much as 50 feet difference in elevation between the high and low points, true levels could not be maintained over the total area of the pit.

Mining of carbonate ore, as stated before, was started in May, 1917, and after about five years of carbonate production, stripping of the sulphide ore was sufficiently advanced to warrant the erection of a concentrating plant. A 5,000-ton concentrator was completed near the close of 1923 and regular production of sulphide ore began in January, 1924. After that time both carbonate and sulphide ore were produced until July, 1930, when the leaching plant was closed and carbonate-ore production ceased. Up to January 1, 1931, 16,800,000

5 - References in this paper to the scale of present operations refer to the period in 1931, prior to September 1.

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tons of carbonate ore and 15,600,000 tons of sulphide ore had been mined by steam shovels from the pit and treated either in the leaching or concentrating plants.

The pit, including the approach, now covers an area of 162 acres, and the bottom level, which is at an altitude of 1,700 feet above sea level, is 317 feet below the top of the highest hill which was mined and 135 feet below the average elevation of the rim. The shape of the pit is roughly elliptical with the major axis pointing northwest-southeast, and is 3,600 feet long by 2,500 feet wide. Four regular levels for the production of sulphides have been opened up, and two additional levels for stripping operations are being maintained.

Entry is made into the pit at its south end with an approach which starts at a point about 2,000 feet from the crushing plants and parallels the east side of the pit at a distance of about 1,000 feet to the east. This approach carries all main tracks connecting with the ore levels. The track in the bottom of the approach is on a 2 per cent adverse grade. Waste as well as ore is hauled through this approach, but is carried over a separate line from the mouth of the approach to the waste dumps. It is planned to connect the upper levels at the south end of the pit with an independent line to the dumps; it is estimated that a large portion of the stripping tonnage can be thus hauled out of the pit without interference with ore haulage.

Stripping

To date the stripping of waste overburden has not been an important factor in the mining of ore. To January 1, 1931, only 6,800,000 tons of waste had been removed in the mining of 32,400,000 tons of ore, a ratio of 0.21 tons of waste to 1 ton of ore. A large portion of this material occurred within the orebody and in the approach, and strictly speaking was not overburden. Stripping will become an important factor when the south end of the orebody, which dips under a barren capping, is uncovered and mined. A waste ratio of about 0.3 ton of waste per ton of ore is being maintained at the present time, and it is estimated that the total remaining open-pit ore as now known will be mined with a ratio of 0.8 ton of waste to each ton of ore. No special equipment is used in the mining and haulage of waste; the steam shovels, dump cars, and locomotives are the same as used for ore production.

The main waste dumps are located on the east side of a low range of hills which lies a short distance east of the pit, where an area of nearly 1 square mile is available for dumping purposes if needed. The material on these dumps contains 0.30 per cent copper, mostly in the form of chalcopyrite.

Mining

The average daily production during the time when both sulphide and carbonate ores were being treated was 9,500 tons —6,500 tons of sulphide and 3,000 tons of carbonate ore. Since July, 1930, only sulphide ore has been mined, and the average daily rate has been about 6,500 tons. The present production, including waste, is obtained in 7 shovel shifts — 3 shovels working during the day shift and 4 during the night shift.

The active mining benches have an average length of about 2,000 feet and an average width of 350 feet. The general practice has been to mine the benches on the east and west sides of the pit separately, and no attempt is made to maintain a bench for the entire distance around the pit. This system evolved naturally from the mining of the carbonates and the opening up of the first two levels in sulphides through an approach which entered the north end of the pit and roughly cut the ore area into two parts, one east and the other west of the approach. After the new approach was connected to the south end of the pit which is at the narrow end of the orebody, the old central approach was mined out and benches on one side

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of the pit were connected with those of corresponding elevation on the other side. The benches are roughly circular at their north ends (fig. 7).

The height of the existing banks ranges from 25 to 35 feet, but 25 feet has been determined as the most economical height, and is being used in opening up new levels. The average slope of the banks is 50° from the horizontal; however, 25-foot banks in average ground will stand nearly vertical if properly trimmed. Occasionally fault planes paralleling the bank and dipping toward the pit determine the bank slope.

The pit has not progressed to the stage where the maximum slope for the sides of the pit can actually be determined; however, it is assumed, for the purpose of making estimates of open-pit ore, that a slope of 45° can be safely maintained. This will be the slope from the rim of the pit to the toe of the bank at the bottom, and will include berms for the protection of the working levels against minor rock slides.

DRILLING AND BLASTING

Each bank in the pit presents its own blasting problem; however, there are definite objects and fundamental limits that are common to all blasts. The principal object of a primary blast is to break as much ground as possible into material which can easily be handled by the shovels; the size of the blast is limited principally by the number of exploders that can safely be detonated at once and by the length of time required to load the holes.

In general, the larger the area covered by the primary blast, the less will be the proportion of material requiring secondary blasting before it can be shoveled, and consequently the cheaper will be the total average cost of breaking the ground. It is advantageous to break as long a section as possible with a single blast, for the reason that there usually is a rib of poorly broken ground between contiguous sections of a bank which have been shot, and consequently the number of these ribs will depend upon the length of the sections blasted. A wide blast, especially in very hard ground, is also desirable for the reason that it gives much better fragmentation than a narrow one, due to the fact that the powder in the center of the shot is more evenly distributed and shatters the ground better than that at either face or bank side of the blast. The face opens up along seams and is thrown forward, and large bowlders, which require secondary blasting, remain on or near the surface. Likewise, a strip of ground along the bank side of the blast breaks along major fractures, and requires secondary blasting before it can be shoveled. The maximum area covered by a single blast will vary from 90 to 100 feet in width and from 200 to 300 feet in length.

The time required for loading the holes is important, especially in wet ground, as in this case there is an appreciable loss in strength of powder from the time it is loaded until it is detonated.

Cushion blasting — that is, blasting behind ground already shot — is seldom used, for the reason that locating holes along a bank that is not faced up is largely guesswork, and unless the rock is quite soft, a poorly broken rib for the entire length of the blast is likely to result from the procedure.

Drilling

It was early demonstrated that air drills were more satisfactory than churn drills for drilling the hard rock encountered in the orebody, especially when the depth of holes did not much exceed 30 feet, and as soon as regular benches could be established, air drills were adopted as standard. The original air drills were piston machines with tripod mountings, and were used successfully until replaced by hammer drills. Drilling tests made in 1923 with hammer drills established the fact that a much greater footage could be obtained with them

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than with piston machines, and several years' experience since then with the hammer type of drill have established its superiority over any other type of drill used. Snce 1923 the piston drills were gradually replaced, and at the present time hammer drills are used exclusively. The average rate of drilling with piston machines was about 24 feet per shift, whereas the average rate with hammer drills is about 80 feet per shift. During a 6-month period extending from November, 1930, to April, 1931, 190,264 feet were drilled in 2,306 drill shifts, or an average of 82.5 feet per shift. No mountings are used with these drills, but they are equipped with spring handles, and when in operation they are suspended from tripods with block and tackle for ease of handling. A tripod consists of three 18-foot pieces of 1-1/2-inch pipe flattened at the upper ends and bolted together with a clevis. The drill is suspended from the clevis with a twofold tackle and can readily be hauled up when it is necessary either to free the steel in the hole or to change steel. An air pressure of 100 pounds per square inch is maintained at the compressors, or between 90 and 95 pounds at the drills. Drill steel is made from 1-1/4-inch hollow-round steel in lengths from 4 feet for starters up to 36 feet for the longest drill with a difference of 2 feet for each change. Cross bits are used. The gage of starter bits is 2-7/16 inches and a decrease in gage of 1/16-inch is made for each change in length.

All holes are drilled 2 feet below grade so that when they are shot no hard ribs or knobs will be left above grade. Spacing of the holes varies with the hardness of the rock and the height of the bank. For a 25-foot bank this spacing ranges from 10 feet in the hardest ground to 20 feet in the softest. Figure 6 shows the arrangement of holes for a standard 25-foot bank; for lower banks the distance between holes would be proportionately less. The distance from the first row of holes to the crest of the bank is less than the distance between succeeding rows, in order to take care of the slope of the bank. Should this slope be very flat, short toe holes are drilled to relieve the burden on the vertical holes. The front row is used as a base, and from it the others are located, the holes in alternate rows being staggered (fig. 6).

A drill crew consists of two machinemen. At the present 18 crews are employed, 9 crews working day shifts and 9 crews on night shifts. Besides the machinemen, 8 muckers are employed on day shift to clear away loose rock around the points where holes are to be drilled and to set casing wherever it is required. A drill foreman lays out and supervises the work of the drill crews and muckers.

All banks are carried on a predetermined grade and grade stakes are set by a level party at 50-foot intervals along a bank which is to be drilled. The drill foreman then marks the position and depth of holes to be drilled, and sets the muckers preparing these sites for the drill crews. In case the ground is fractured from previous blasts, short lengths of 2-1/2-inch pipe are set for casing the collar of the hole. When the section of the bank which is to be shot in one blast has been drilled, drilling is continued into the adjacent section, but a strip of undrilled ground is left between the two. This prevents loss of holes when the first section is blasted, and additional holes can be drilled afterwards, if necessary, and shot with the next blast.

Blasting

All holes are sprung with 1-1/8-inch 50 per cent strength gelatin stick powder; 40 per cent strength granular powder, known as Quarry Special No. 1, is used for the main charges in dry ground, and 1-1/8-inch 40 per cent strength gelatin stick explosive in wet ground. Forty per cent strength gelatin stick powder is also used for the main charges in toe holes. Granular powder is fairly free running and has an advantage over stick powder in that it can be loaded much faster.

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Primers are made of a single stick of 1-1/8-inch 50 per cent strength gelatin powder in which one No. 8 electric detonator is inserted. The detonator is inserted in one end of the stick, and the lead wires brought up along the sides of the primer to the end opposite the detonator, where a half-hitch is taken with each wire. The hitches are made so that the free ends of the wire will be on opposite sides of the primer and it will hang vertically with the cap at the lower end when lowered into the hole. After the hole is about 80 per cent loaded, the primer is lowered and pressed into the charge, and the remainder of the charge is then placed on top of the primer, thus insuring direct contact of the primer with the main charge. Two primers are used in holes in ravelly ground, one placed at the bottom of the pocket and the other at the top of the charge.

Pockets for the main charge are made at the bottom of the drill holes by springing them with 50 per cent strength gelatin stick powder. In this process the hole is deepened a few inches, as it is best to have the pocket well below grade. The number of springings required to form a pocket of sufficient size to hold the charge varies with the hardness of the ground. Two springings usually are sufficient, but in some ground as many as six are necessary. The ratio of the amount of powder used in any springing charge to that necessary to fill the pocket after it is sprung is fairly constant for a particular hole, so that by determining this ratio for the first springing it can readily be calculated how many times the hole need be sprung in order to hold the final charge. If, for instance, two sticks of powder used in the first springing makes a pocket large enough for 10 sticks of powder, the ratio is 1 to 5, and the second springing will then make a pocket large enough for 50 sticks. If the load for the hole is equivalent to 50 sticks or less, two springings will, therefore, be sufficient; if the load is more, additional springing will be necessary.

The first springing charge, which usually is 2 sticks of powder, is stemmed with tailings sand for a distance of about 2 feet above the charge. After shooting, the hole is blown out by compressed air. Succeeding springing charges are stemmed with greater amounts of sand Springing is carried on in rotation over as large an area as practicable, so that after blowing out the holes, sufficient time has elapsed for the holes first sprung to be cool enough for another charge. All of the holes of the blast are sprung before the final loading of the holes is started.

Granular powder, which is used for the main charge, is poured directly in the holes and lightly tamped with wooden tamping sticks to insure filling the pockets. After the primers have been placed in the charge and the pockets have been filled with powder the holes are stemmed to the top with dirt. In very hard ground, where the holes are more than 20 feet deep, an auxiliary load consisting of from 10 to 20 sticks of 50 per cent gelatin powder is placed about halfway up the hole to insure thorough breaking of the rock near the surface. Wherever this auxiliary load is used, two primers are always placed in the main charge and one primer in the auxiliary charge. When all holes are loaded, lead wires of No. 10 insulated copper wire are strung between the holes and tested. The exploders are then connected in parallel to the lead wires, and the blast fired by current from a 110-volt power line; the firing switch is located near the main line at a safe distance from the blast. After the bank has been shot, it is inspected, and loose casings and wires are removed.

The total amount of powder used in a primary blast varies with the hardness of the rock and the burden on the blast. As a guide for loading, 1/2-pound of powder is used per ton for hard rock, 1/3-pound per ton for medium rock, and 1/4-pound per ton for soft rock. This is a slight overload, but with this as a base and with some experience the right amount can be determined for each class of rock. The calculated loads do not take into consideration any overbreak, and as there is always a large tonnage broken beyond theoretical lines, the actual tonnage broken per pound of powder will be greater than that calculated. Due to improved blasting practice and lower banks, the amount of material broken per pound of powder has

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shown a decided increase during the past 10 years of mining as shown in Table 1. No separate account is made of powder used in primary and secondary blasting, and the above figures are on the basis of total powder consumption.

As the spacing of the holes drilled for blasting is determined by the hardness of the rock, the amount of powder used in any hole will depend principally on its depth. About one box or 50 pounds of powder is used as the main charge in a 20-foot hole; 1-3/4 to 2 boxes in a 25-foot hole; and 3-1/4 to 3-3/4 boxes in a 30-foot hole. The charges in the row of holes nearest the face of the bank vary somewhat due to irregularity of the toe, but all back holes of a blast are loaded with the same amounts of powder.

The material broken by a primary blast is comparatively coarse, as stated before, because of the presence of widely spaced fractures in the rock formation; roughly one-half of the broken ore will be in pieces larger than 18-inch cubes. Before the ore can be loaded into cars it must be broken to a size to pass the shovel dipper, which has a clearance of 4 feet by 3 feet. Bailing oversize bowlders into the cars is not permissible for the reason that the primary crushers will handle no larger pieces of ore than will pass through the dipper. One jackhammer man at each shovel blockholes the large bowlders as they are encountered in the shoveling of the ore. A hole is drilled with a light jackhammer to the center of the bowlder, and blasted with from 1/4 to 1/2 stick of 40 per cent gelatin powder.

Table 1.- Tons of all material (ore and waste) broken per pound of powder consumed

Year	Dry tons per pound
1921	3.05
1922	3.27
1923	2.67
1924	2.27
1925	3.37
1926	3.93
1927	4.22
1928	4.39
1929	5.85
1930	5.93

LOADING THE ORE AND WASTE

Ore and waste are loaded by means of 105-ton railroad-type steam shovels fueled with oil and equipped with 4-yard dippers, 20-foot 6-inch dipper sticks, and 30-foot booms. Ten of these shovels are now available for operation. This type has been used since the beginning of shovel operations, and although many minor changes in the design of the shovels have been made from time to time to adapt them to the particular conditions at Ajo, the general specifications of all shovels are the same. The fact that the first shovel placed in operation is as serviceable now as it was originally, after 14 years of almost continuous use, is evidence of their ruggedness as well as the care with which they have been serviced and repaired.

Waste is shoveled with the same equipment as is used for ore, and the only difference in the two operations is that the waste trains are routed to the waste dumps instead of to the crushers.

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From a study of the air-drill assays, the mine superintendent designates any portion of banks which are to be wasted. The method of ore control is more fully described under "Sampling." By this method a close separation of ore and waste is easily made, especially when shoveling in a bank not exceeding 25 feet in height.

The shovel crew consists of one runner, one craneman, one fireman, and four pitmen. One of the pitmen is also a jackhammer man and does all blockholing of bowlders in front of the shovel and hard nobs in the bottom of the cut.

The maximum width from which a shovel can load is 48 feet, measured from the inside rail of the loading track to the toe of the cut. This is equivalent to a cut of approximately 30 feet in the solid bank. The maximum height at which the dipper can dump its load is 18-1/2 feet above the section rails, which limits the depth of drop cuts to about 10 feet.

The shovels move on 6-foot sections of track made of 70-pound rails; the shovel track is laid parallel and at a distance of 23 feet from the loading track. When the end of a cut is reached, the shovel is moved back on the loading track to a new position, and the loading track is either lined over to the blasted bank, or is torn up and relaid. The latter method usually is employed, since it saves considerable grading and gives a better track. Some track is laid in panels with a locomotive crane, but this method is never as satisfactory as relaid tracks with staggered joints.

A new level is established by making successive drop cuts from the old level until the required elevation has been reached. The first drop is carried on a 2 per cent down grade until a depth of 10 feet has been reached, from which point it is carried forward 10 feet below the original level. When this cut has been finished, the loading track is laid in its bottom and the second cut is then made in either bank. The second cut is made in the same manner as the first, excepting that the grade is continued downward until a depth of 10 feet below the new loading track elevation has been reached. Thus in establishing a level 25 feet below the old level, three drop cuts are necessary.

Fueling of shovels is accomplished by attaching a 3-inch hose to the oil tank on the locomotive tender by special flange connection and pumping oil from the locomotive to the fuel-oil tank on the shovel. The shovel tank has a capacity for 1-1/2 shifts, and it usually takes from 5 to 10 minutes to refuel. The tank on each locomotive has a capacity sufficient to refuel one shovel and in addition to run the locomotive for an 8-hour shift.

Minor repairs to shovels are made by the operating crews between trains; heavy repairs and moving of shovels are handled by the repair crew. Shovels usually are washed out once a month by the repair crew and are completely overhauled about once every five years.

During the year 1930 a total of 3,410,114 dry tons of material was loaded in 2,580 shovel-shifts, or an average rate of 1,322 dry tons per shovel-shift. Since the beginning of operations the maximum amount of material handled by any shovel during a shift was 107 cars, or approximately 4,200 tons.

TRANSPORTATION

The transportation equipment for hauling ore and waste from the pit consists of 15 oil-fired steam locomotives and 82, 20-yard, automatic, air-dump cars. The locomotives are class 060 S134. The weight of a locomotive alone is 134,000 pounds, and that of the tender 88,800 pounds. The maximum tractive power is 28,700 pounds. The tender is 8-wheeled and has a capacity of 4,000 gallons of water and 1,500 gallons of oil. The locomotive boiler is designed for 180 pounds steam pressure.

The dump cars are flat-bottomed and can be dumped from either side. The over-all length of a car is 32 feet, and outside width of the body is 10 feet 2 inches. The height from the top of the rail to the top of the car is 8 feet 5-1/2 inches. Five or six cars normally constitute a train, the average load for a car being 40 tons of ore or waste.

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The present track system from pit to crushers and dumps has a total length of 13 miles of standard-gage track, 10 miles of which is in the pit and approach to the pit, 2 miles on the waste dump and approach, and 1 mile on the main line from the pit approach to the crushing plants. The maximum adverse grade on tracks is 2 per cent, compensated on curves, and the maximum curvature on main lines is 20°. All tracks are laid with 70-pound rails and 8 by 8 inch by 8-foot cross ties. Most of the shifting of tracks is done with a track shifter. The track gang consists of 25 men, supervised by a foreman and two subforemen.

Trains are hauled from the shovels, where they are loaded, directly to the crushing plants or dumps. The average length of haul to the crushing plants is 1.5 miles, and to the waste dumps 1.8 miles.

DRAINAGE

A system of dams and ditches diverts rainfall run-off away from the pit, so that only water that actually falls within the pit area need be handled. The lowest level in the pit has a slight slope to a central low point, where an air-operated pump with a capacity of 75 gallons per minute lifts the water over the edge of the pit to the outside drainage system. As the average rainfall at Ajo is under 10 inches annually, very little inconvenience is caused by water collecting in the pit, excepting at infrequent times when very heavy rains occur.

LIGHTING

The shovels are equipped with electric lights for night work. The current for the shovel lights, as well as lights for the drill crews, is obtained from the main power line into the pit, which also supplies current for blasting.

PLANTS

A 15,000-kilowatt steam turbine electric power plant, near the crushing plants and machine shop, furnishes all electric power for the mine, plant, and townsite. The mine uses less than 5 per cent of the total power generated. The compressor and drill-sharpening shop uses 60 per cent of the electric power consumed at the mine, and the remainder is consumed mainly by lighting. A 2,300-volt line connects the compressor plant with the power house.

Water for use at the mine, plant, and townsite is pumped from a well 7 miles north of Ajo to storage tanks above the concentrator. A 6-inch pipe line carries water from the storage tanks to the mine.

The compressor plant, drill-sharpening shop, and shop for minor repairs are at the mouth of the approach to the pit in close proximity to the main tracks.

The compressor equipment consists of three angle compound compressors having a total capacity of 6,200 cubic feet of free air per minute. Each compressor is composed of two independent 2-stage units direct-connected with a synchronous motor. A pressure of 100 pounds per square inch is maintained in the air receivers at the compressor plant.

The drill-sharpening shop is equipped with two sharpeners, one for sharpening the primary drilling steel, and the other for the secondary or plugger steel, and three oil furnaces, two for heating steel and one for tempering. Sharpened steel is delivered to the working places and dull steel returned to the shop in a specially designed flat car which is hauled by a gasoline-operated track shifter. The track shifter is in use for about 8 hours each day for delivering steel and powder; the balance of the time it is available for track shifting.

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The powder magazine is located near the waste dump approach track at a safe distance from the plant and mine. Eighteen-inch adobe walls and a 4-inch layer of sand over the ceiling makes the powder house practically bulletproof.

PER CENT OF EXTRACTION

The total tonnage of ore mined and treated to date is over 5 per cent in excess of that estimated from diamond-drilling data with no appreciable difference in the average grades of ore as estimated and treated.

WAGE, CONTRACT, AND BONUS SYSTEMS

All labor is paid according to a daily wage system, and no bonus or contract system is in effect. The following tabulation shows the average wage scale for 1930 for labor at the mine:

Shovel runners	$8.15
Cranemen	5.70
Powdermen	4.66
Machine drill runners	4.48
Firemen	4.37
Jackhammer runners and powdermen helpers	3.14
Pitmen	2.74
Trackmen and muckers	2.56

Table 2.- Summary of costs

		Year 1930
Ore loaded during period	dry tons	2,376,764
Waste loaded during period	do.	1,040,610
All material loaded during period	do.	3,417,374

Operating costs per dry ton of all material mined

	(1)	(2)	(3)	(4)	(5)	(6)	(7)	(8)
Drilling and blasting	$0.035	$0.003	$0.001	$0.003	$0.001	$0.025	$0.002	$0.070
Loading	.026	.001	-	.001	.013	-	.008	.049
Transportation	.042	.001	-	-	.016	-	.007	.066
Miscellaneous	.001	.016	-	-	-	-	.008	.025
Total	0.104	0.021	0.001	0.004	0.030	0.025	0.025	0.210

(1) = Labor
(2) = Supervision
(3) = Compressed air, drills, and steel
(4) = Electric power
(5) = Fuel
(6) = Explosives
(7) = Other supplies
(8) = Total

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GENERAL COST DATA

Labor:		
	Total man-shifts for year	80,994
	Dry tons, all material mined per man-shift	42.19
	Man-hours per dry ton, all material	0.19
Explosives:		
	Pounds of powder consumed during year	576,150
	Dry tons, all material mined per pound of powder	5.93
Power:		
	Total kw.h. used during the year	1,903,947
	Dry tons all material mined per kw. h.	1.79
Fuel:		
	Barrels of fuel oil consumed by steam shovels during year	26,956
	Barrels of fuel oil consumed by locomotives during year	28,387
	Total barrels fuel oil consumed	55,343
	Dry tons all material mined per barrel fuel oil	61.75

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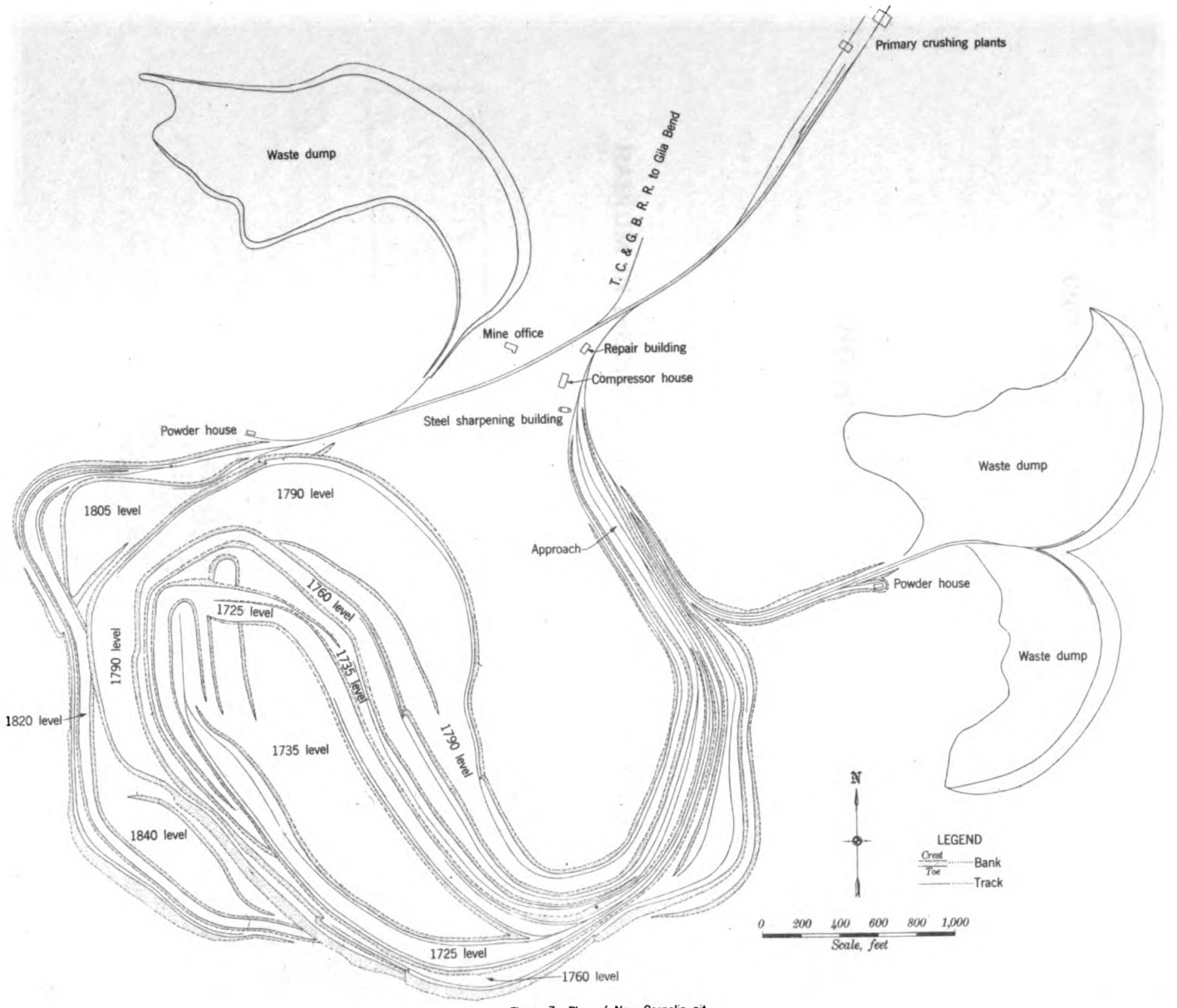

Figure 7.—Plan of New Cornelia pit

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